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UNITED STATES BUREAU OF MINES
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MILLING PRACTICE OF THE KIRKLAND LAKE GOLD MINES (LTD.), KIRKLAND LAKE, ONTARIO

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JOHN DIXON

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MILLING PRACTICE OF THE KIRKLAND LAKE GOLD MINES (LTD.), KIRKLAND LAKE, ONTARIO[1]

By John Dixon[2]

INTRODUCTION AND ACKNOWLEDGMENT

This paper is one of a series on milling methods and costs being published by the Bureau of Mines.

Acknowledgment is made to J. B. Tyrrell, managing director, and Wm. Sixt, manager of the Kirkland Lake Gold Mines, for permission to publish this article.

The Kirkland Lake Gold Mining Co.'s mill is located on Government Road at the west end of the productive part of the Kirkland Lake mining area, 1 mile west of the village of Kirkland Lake. The crushing plant is connected to the shaft house of No. 2 shaft, which is now the only operating shaft.

The mill, which stands about 60 feet from the shaft house and midway between it and the Government Road, to the south, has an average capacity of 160 tons per day and consists of one unit. Grinding is done in cyanide solution by an 8-foot by 30-inch Hardinge ball mill and a 5½-inch by 16-foot tube mill. The cyanide plant employs counter current decantation in Dorr thickeners, followed by filtration on an Oliver filter.

Water for milling and tailings disposal is mine drainage supplemented by water purchased from the municipal plant of the village of Kirkland Lake. Hydroelectric power is purchased from the Northern Ontario Light and Power Co., a subsidiary of the Canada Northern Power Co.

ORE TREATED

The ore at present carries about $13 in gold. A little silver is present and is paid for by the mint, but is of no commercial importance. The ore is altered and silicified country rock. The country rocks in the mine are lamprophyre, quartz porphyry, syenite, and diabase, and any of these rocks may be altered and impregnated with gold up to ore grade.

The gold is very finely disseminated and some is probably in the form of gold telluride, although this has never been definitely proved. Lead telluride has been recognized, and tellurium can always be found in the furnace products from the refinery. Gold-bearing pyrite accounts for some of the gold. Values are not entirely unlocked for cyanidation or concentration at 300 mesh, although that seems about the economic limit of grinding at present.

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[1] - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6508."

[2] - One of the consulting engineers, U. S. Bureau of Mines, and mill superintendent, Kirkland Lake Gold Mines, Ltd., Kirkland Lake, Ontario, Canada.

Gangue minerals are acid and basic feldspars, augite, hornblende, carbonates, and quartz. Physically the ore is generally tough, hard, and close-grained, without cleavage planes or other lines of weakness.

Mill feed carries 2 per cent of moisture.

There is no change, physical or chemical, in the broken ore lying in the stopes.

HISTORY OF PLANT OPERATIONS

The mill started operating in April, 1919, as a counter-current decantation plant treating $5 or $6 ore. It was closed during the summer of 1919, owing to a miners' strike. It was reopened in November, 1919, and operated continuously until January, 1924. During this period no radical changes or additions were made in the mill. The grade of ore varied from $6 to $10 per ton, and extraction was from 85 to 90 per cent. Grinding was about 70 to 85 per cent minus 200 mesh. The mill was closed from January, 1924, till October, 1926, due to the exhaustion of known ore bodies. Milling was resumed in October, 1926. As ore from lower levels was treated, the grade became higher and the gold more difficult to dissolve. Finer and finer grinding became necessary to maintain the percentage of extraction, and the absolute value of the tailings went up with grade of ore.

In 1927, 1½-inch cast-iron balls were substituted for flint pebbles in the tube mill. The change was made without trouble by feeding the metal instead of flint and using a mixture while the flint charge was grinding out. The tube mill is now operating with a low ball charge without trouble, and its capacity could be increased from 50 to 75 per cent by increasing the charge of balls.

In February, 1930, an 8 by 8-foot classifying cone was put in closed circuit with No. 1 thickener, No. 1 agitator, and the tube mill. In July, 1930, this was replaced by a 16 by 4-foot Dorr thickener running as a classifier, as shown in the flow sheet (fig. 1). The underflow from this machine is pumped by a 4-inch diaphragm slime pump directly to the feed box of the tube mill, and the overflow goes to the secondary agitation and then to the decantation system.

With finer grinding, colloids began to give trouble by trapping high-grade solution in floccules of coagulated slime in such a way that it could not be separated by decantation. This problem was met in 1928 by installing an Oliver filter to be used after the last step of decantation, as shown in the flow sheet.

In April, 1927, Crowe vacuum apparatus was installed. This resulted in saving about two-thirds of the zinc-dust formerly required for precipitation, as well as giving smoother plant operation and higher-grade precipitates.

The various changes mentioned have resulted in slightly raising the percentage of extraction without increase of costs, in spite of higher grade and more refractory ore.

METHOD OF MILLING

Figure 1 is the flow sheet of the mill. Figure 2 is a diagram showing the end elevation of the mill. The flow sheet shows roughly, in plan, the course of the ore through

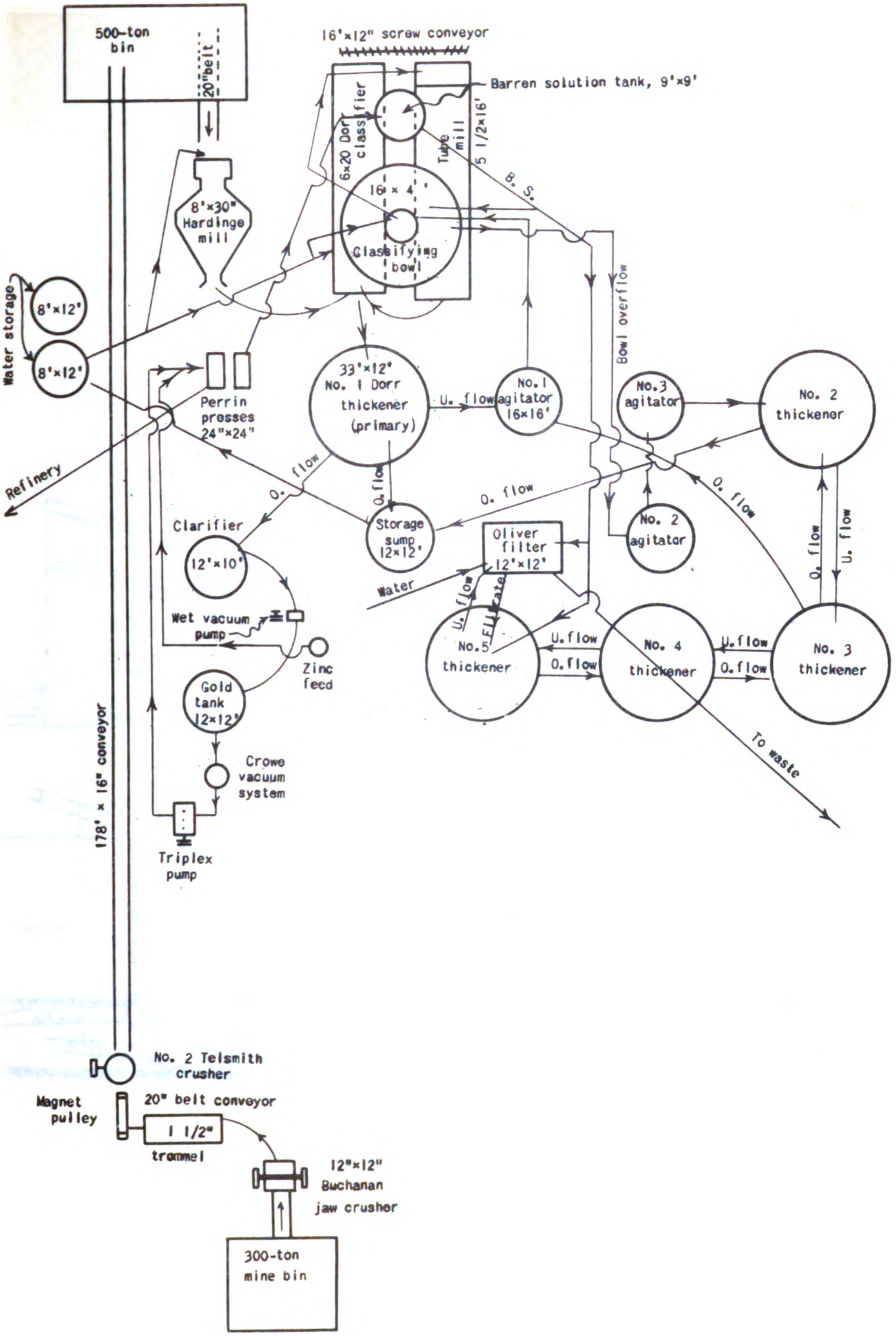

Figure 1.- Mill flow sheet

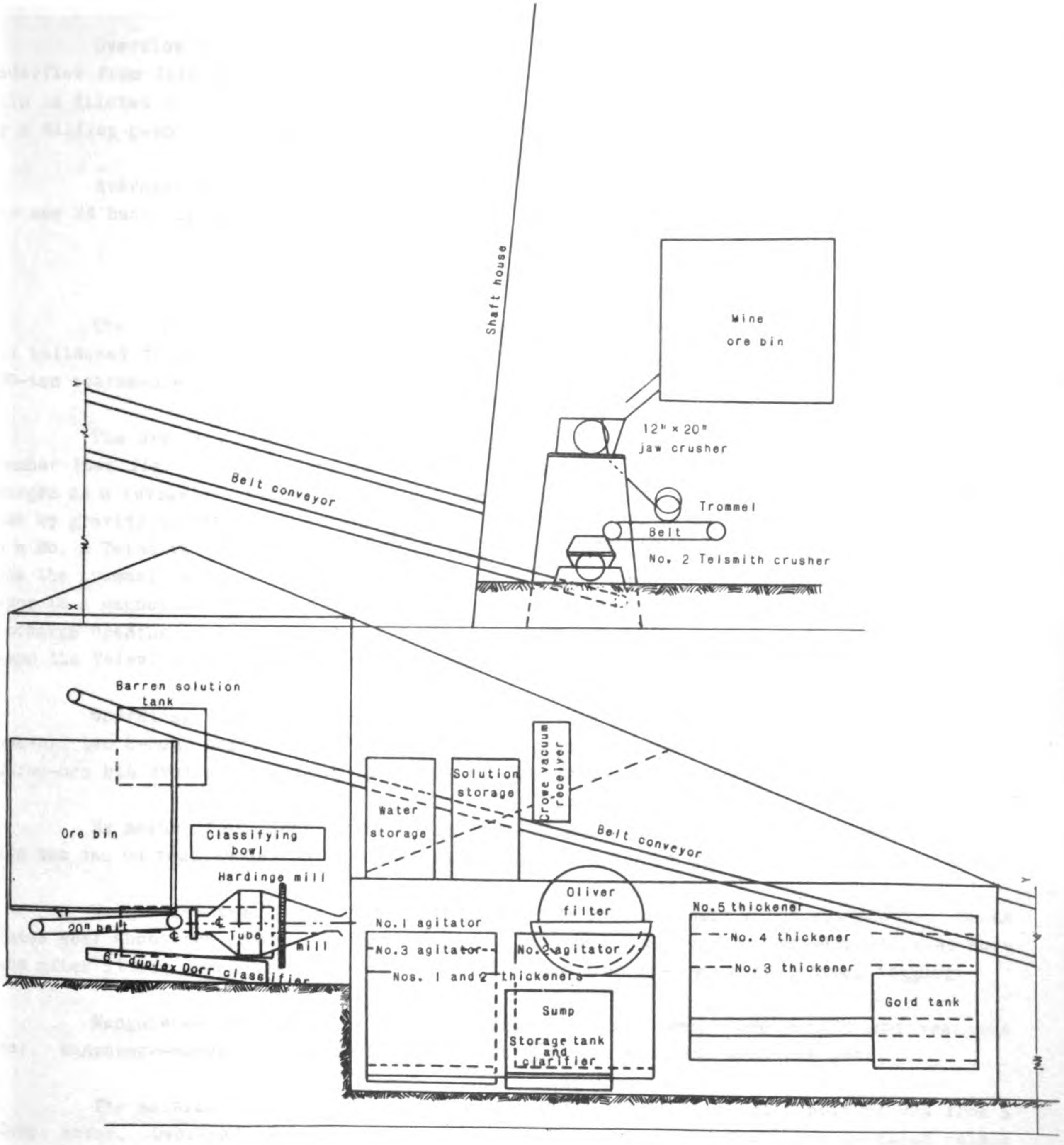

Figure 2.- End elevation of mill

the mill. The travel in the cyanide plant is at right angles to the course through the grinding plant. This results in a compact mill, but does not lend itself to making the present installation a unit in a larger mill.

Overflow from the duplex Dorr classifier goes by gravity to No. 1 thickener. The underflow from this primary thickener is pumped to No. 1 agitator. In No. 1 agitator the pulp is diluted to 70 per cent moisture by the overflow from No. 3 thickener and then pumped by a Wilfley pump to the bowl mounted above the tube mill and No. 1 Classifier.

Average tonnage for the whole mill for 24 hours is 142 tons. The greatest tonnage for any 24 hours is 200 tons.

Breaking

The ore is raised from the mine in 1-ton cars and dumped on a bulldozing platform and bulldozed through grizzly bars, spaced 6 inches apart by spacers 3 feet apart, into a 300-ton coarse-ore bin.

The ore from the coarse-ore bin is fed by gravity to a 12 by 20 inch Buchanan jaw crusher (see fig. 2). This crusher is set with about a 3-inch discharge opening and discharges on a revolving trommel 3 feet in diameter with 1½-inch round openings. The undersize goes by gravity to the main conveyor, and the oversize is delivered by a short belt conveyor to a No. 2 Telsmith crusher set to deliver a product equivalent in fineness to the undersize from the trommel, which it joins on the main conveyor. The head pulley of the oversize conveyor is a magnetic pulley to remove tramp steel from the feed to the Telsmith crusher. The discharge opening of the jaw crusher is so adjusted that the oversize from the trommel just keeps the Telsmith crusher running to capacity.

Operating capacity of the crushing plant as a unit is about 20 tons per hour. However, two 8-hour shifts are often used on the crushers because of lack of capacity in the coarse-ore bin available without resorting to hand shoveling for unloading the bin.

By means of an electric signaling system, the crusherman, ball-mill man, and solution man can be readily called to assist each other in any part of the mill or crusher house.

Manganese-steel jaw plates weigh 420 pounds each and wear about six months. Cheek plates wear about two years. Cast-iron toggles wear about one year. Cast-iron toggles were used after it was found that a great deal of breakage resulted with cast-steel toggles.

Manganese-steel mantles for the Telsmith crusher weigh 560 pounds and wear one year. Manganese-steel concaves weigh 470 pounds per set and also wear one year.

The machines are belt driven from a countershaft which is also belt driven from a 40-hp. motor. Overloads are taken care of by belt slippage, and fuses and overload relays on the motor.

Primary Grinding

Primary grinding is done in one 8 foot diameter by 30-inch Hardinge ball mill running at 16 r.p.m. The rated speed of this mill is 21 r.p.m., but as the full capacity of the mill is not required, the mill is run at the reduced speed with reduced capacity. This mill runs in an open circuit with no circulating load. However, there is a 12-inch diameter by 3-foot trommel with 3/8-inch round openings bolted to the discharge bell of the mill. About a ton per day of tramp oversize is removed from the mill discharge and returned to the mill by hand. The ball mill discharge averages 75 per cent solids.

The average hourly capacity in 1930 was 6.73 tons, and the ball mill operated 89 per cent of the time. Both the running time and capacity of the mills are limited by the settling capacity of the decantation plant. Table 1 shows screen analyses of the mill products.

Five-inch forged-steel balls are used, manufactured by the Hull Iron and Steel Co. of Hull, Canada. Ball charge is about 30,000 pounds.

Hardinge plate and wedge liners of manganese steel are used. Ball consumption averages 1.96 pounds per ton of ore. Liner consumption averages .24 pound per ton of ore.

The mill is belt driven at 30-foot centers through a spur gear with a gear ratio of 5 to 1, from a 125-hp. squirrel-cage motor. Starting overload is taken care of by a Smith-type Hill friction clutch.

Secondary Grinding

Secondary grinding is done in one 5½-foot diameter by 16-foot power and mining tube mill running at 28 r.p.m. This mill runs in closed circuit with 6 by 20-foot Dorr classifier, and also with a classifying bowl, as shown in the flow sheet and mentioned in the historical review.

The total feed to the tube mill is 12.9 tons per hour. Of this, 6 tons per hour is underflow from the bowl and 6.9 tons per hour return sands from classifier. Assuming that 90 per cent of the total ball-mill feed goes through the tube mill, the circulating load is 2.15 times its original feed.

The tube-mill discharge pulp is kept at about 73 per cent solids; the classifier overflow at about 25 per cent, and the bowl overflow at about 18 per cent solids.

One and one-half inch balls are used, forged from the tops of old railroad rails, by the Burlington Steel Co. of Hamilton, Ontario. The mill is designed for a ball load of 30,000 pounds. The actual ball load has not been determined, but it is probably about half loaded. Ball consumption averages 2.5 pounds per ton. Liner consumption averages .26 pound per ton.

A modified El Oro type of chilled white iron liner is used, manufactured by the Cobalt Foundry of Cobalt, Ontario. Figure 3 shows a drawing of this liner.

The mill is belt driven by a 100-hp. squirrel-cage motor at 30-foot centers through a spur gear with a gear ratio of 5 to 1. Starting overloads are taken care of by a Smith-type, Nill friction clutch.

The Dorr classifier has a grade of 2-1/8 inches per foot and makes 12 strokes per minute.

Cyanidation

The flow sheet of the cyanide plant is shown in Figure 1. The method in use is all sliming in cyanide solution and continuous countercurrent decantation followed by filtration. This system is a development due to changing conditions and the progress of the art. This has been discussed quite fully in the historical review.

An immense amount of research has been done in the effort to devise a more effective treatment, both by ourselves and by other mines in the district. Flotation (either of raw ores or tailings) has not yet been applied with commercial success. Extremely fine grinding is necessary to free values even for flotation, and after the concentrates are made, the commercial recovery of the gold from the concentrates presents a difficult problem. We have tried adding fresh cyanide to the ball mill, to the tube mill, to the primary agitator, and to the secondary agitator. It makes no measurable difference where the cyanide is added as long as the strength is maintained. We now add fresh cyanide to No. 1 agitator.

Grinding solution is kept at 0.9 pound. NaCN equivalent and the other solutions stay at about 0.7 pound per ton. Increasing the strength of cyanide increases both mechanical and chemical loss without affecting the extraction. Decreasing cyanide strength does not necessarily decrease extraction, but when the attempt has been made to carry a lower cyanide strength, it has occasionally resulted in high tailings assays. While it is not certain that the assays were affected by low cyanide, the saving of cyanide by using a strength below 0.9 pound per ton of solution was so small as not to be worth while.

Lime is added at the ball mill and all solutions are saturated with $Ca(OH)_2$. The high lime is necessary for maximum rate of settling and presumably helps to break down tellurides.

No. 1 thickener underflow averages about 58 per cent solids. This is much heavier than the other thickener underflows because, as seen from the flow sheet, the No. 1 thickener has a large, comparatively coarse circulating load. No. 1 agitator is carried at 25 per cent solids and like No. 1 thickener carries a heavy circulating load. This results in giving a preferential agitation to the heavier and higher-grade particles of the ore.

The other agitators are kept at about 18 per cent solids, and the other thickener underflows at about 45 per cent solids.

The total time of treatment is indefinite, due to the preferential treatment and variation in the settling rate, but a change in grade of ore is usually reflected in the tailings after about 36 hours.

The displacement of solution in the thickeners, which are of continuous, rim-launder type, is about 2 tons of solution per ton of ore. The solution displacement on the filter is about 1/2 ton of solution per ton of solids.

Air is introduced incidentally in the air lifts of the Dorr agitators but they are not at all effective as aerators. The solution pumped from the storage pump to the upper storage tank is raised 12 feet above the tank and allowed to plunge into it. Oxygen tests show that this is a very effective way of introducing oxygen. Kieselguhr diffusers are placed in the barren solution tank, in the storage tank, and in the overflows from the thickeners. These are fed with compressed air at about 5 pounds pressure, and are very effective aerators. Sodium peroxide and barium peroxide have been tried in various ways and at various times, but with no effect.

Butters-type filter leaves are used to clarify the pregnant solution for precipitation.

The tailings are filtered by one 12 by 12 foot Oliver drum filter. In the filtering cycle 1/4 of the time is used in loading, 3/10 in barren wash, 1/5 in water wash, 1/12 in blow-off and 1/6 is blank. About 0.3 ton barren wash and about 0.2 ton water wash are used per ton of solids. Blow-off air is taken from the mill compressor through a reducing valve at 3 pounds pressure. Vacuum is maintained at 23 to 24 inches of mercury. Twenty-ounce cotton twill is the filtering medium, and the capacity averages 700 pounds of solids per day per square foot of filtering medium with the thin pulp fed to the filter. Cake is discharged containing 80 per cent of solids.

Analysis of pregnant solution is shown in Table 4.

Pumping

Pumping is not a serious problem and has been mentioned incidentally in previous discussions. Slime pumping is done with Barnes 4-inch diaphragm pumps for which we have designed our own valves. The precipitation pump is an Aldrich 5 by 6 inch triplex pump of 100 gallons per minute capacity. Other solution pumps are small centrifugals. Vacuum for the clarifier leaves is furnished by an 8 by 10 inch Gould geared wet vacuum pump. Vacuum for the Oliver filter is supplied by an Oliver type M. S. 14 by 8 inch vacuum pump. The centrifugal pump taking filtrate from the filter has a gland especially designed by the Oliver company to maintain suction against a high vacuum. A 6 by 4 inch single-cylinder, high-speed, Ingersol Rand vacuum pump maintains 26 inches of vacuum for the Crowe vacuum system.

Precipitation

Gold is precipitated from the clarified, deaerated pregnant solution by means of zinc-dust. The solution averages about $4 per ton in gold. Zinc-dust is fed by means of a slow moving 3/4-inch auger in the bottom of a hopper. The zinc drops into a cone which is kept full of barren solution, the supply of which is regulated by means of a float valve. The emulsion of barren solution and zinc is pumped from the bottom of the cone by a 12½ by 4 inch Gould single-cylinder geared pump to a small receiver near the presses where it joins the pregnant solution from the precipitation pump.

Originally the zinc was fed into the suction of the precipitation pump, but after the installation of the Crowe vacuum system, precipitates lodged in passages of the pump and in the pipe lines and were very difficult and expensive to dislodge; they caused difficulty in checking recovery and occasioned loss of precipitates in cleaning pipe lines.

Two square Perrin plate and frame filter presses, 24 by 24 inches in size are used alternately, dressed with 10-ounce duck covered by filter paper. In normal operations except for three hours at the start of a new press the barren solution contains only a trace of gold. Ninety-nine per cent of the gold in the pregnant solution is precipitated.

During 1930, 1.9 ounces of zinc was used per ounce of gold and silver recovered.

Lead nitrate is fed to No. 1 agitator at the rate of 1½ pounds per day. This prevents fouling of solutions and promotes good precipitation. Although precipitation may remain good for months at a time, if no lead is used, yet when for any reason precipitation does become bad, it is a great deal worse if there is no lead in the solutions.

Refining

Eighteen hours before a press is to be cleaned up, the solution is turned off and compressed air is turned into the press. After blowing for 18 hours the air is turned off and the press opened and cleaned. The canvas is left to use again and the paper separated from the precipitates and burned separately in the furnace. The long blowing with air oxidizes a great deal of the excess zinc left in the press and leaves the precipitates quite dry.

In the refinery the precipitates are mixed with flux by shoveling over and over in a steel mixing box. The following charge is used:

	Pounds
Precipitates..........	100
Manganese dioxide	4
Ferric oxide..........	4
Crude soda..............	6
Air-slake lime......	16
Sodium nitrate......	10
Borax glass............	25
Silica sand............	25

The thoroughly mixed charge is shoveled into an oil-fired Rockwell tilting furnace 2 feet 6 inches in diameter by 3 feet 6 inches long, outside dimensions. The furnace has a rammed carborundum lining, sold under the trade name of "carbofrax." We drilled a number of 3/4 inch holes in the steel shell of this furnace to make it easier to break out an old worn-out lining from the outside.

After a charge comes to a quiet fusion fresh charge is added and the furnace is heated again; the operation is repeated until the furnace is full of molten slag. Slag is then poured into a conical slag pot, and a fresh charge is added. This procedure is continued until the precipitates are all melted. Most of the slag is then poured into slag pots, after which the gold is also poured into slag pots, care being taken that the metal buttons are not too large to handle and to charge back into the furnace.

The buttons, when cool, are freed from adhering slag and charged back into the furnace and melted. Sodium nitrate is thrown on the pool of molten metal. This reacts with the base metals, forming fumes and a crust of metallic oxides. The crust is skimmed off and the metal is heated again. This procedure is repeated until bullion of the desired fineness is indicated by the metal hardening at a high temperature. The bullion is then poured into bullion molds for shipment.

Skimmings and slag from the buttons and bricks are added to the next clean-up. The rest of the slag is returned to the ball mill. Investigation has shown that all the gold in the slag is easily dissolved in the mill and that retreating the slag in the mill does not foul the solutions. When a lining is removed, any coarse pieces of gold are picked cut by hand and added to the next melt. The scrap lining is thrown into the ball mill. The gold in the lining is easily dissolved and the lining material has no precipitating effect on the solutions. As lining and slag are impossible to sample accurately, the refinery receives no credit for the gold thus returned to the mill. There is a small but undetermined loss of gold in fumes; and there are no other refinery losses except the mechanical loss that is unavoidable when handling such high-grade material. The bullion produced varies from 930 to 955 parts in a thousand total fineness of gold and silver. Precipitates are usually about 50 per cent fine gold and silver.

Launders

Pulp launders throughout the mill have a grade of 1 inch per foot. The discharge launders from the ball mill and from the tube mill are of wood-lined with steel plate. All other launders are wood, unlined.

Tailings Disposal

Tailings are diluted with water to a ratio, water-to-solids, of 4 to 1 and are pumped by a Wilfley 3-inch sand pump through 6-inch, inside diameter, wood stave pipe, 2,800 feet to Kirkland Lake. The pipe line is laid out carefully on a grade of 1/4 inch to the foot and supported on trestles. Extreme care was taken to leave no low places, so that the pipe drains itself and does not freeze up in cold weather when the flow stops for any reason.

Sampling and Estimating Recovery

Head sample.- At 2-hour intervals a cut is taken from the ball feed belt, 1 foot in length and across the belt. This ore is weighed and returned to the mill. The revolutions of the head pulley of the conveyor are counted by an automatic counter. The average of the "belt weights" multiplied by the revolutions of the head pulley and multiplied by a constant equals the tons of ore fed to the mill. The constant includes the circumference of the head pulley, the thickness of the belt, the conversion of pounds to tons, and a correction for the average moisture of the ore.

At 2-hour intervals (all samples are taken by hand and at 2-hour intervals by cutting the full pulp stream with a sampling dipper, which is emptied into a sample pail) a sample is taken from the discharge of the ball mill. At the same time a specific-gravity determination is made of the ball-mill discharge and a sample is taken of the storage solution entering the ball mill. The ball-mill discharge sample at the end of the mill day is thoroughly mixed and quartered down wet. The quartered down sample weighing about 250 grams is filtered and washed with water, making about 600 c.c. of filtrate. The filtered pulp is weighed, dried, and assayed. The filtrate is measured and assayed.

The dissolved gold in the ball-mill discharge equals

$$\frac{\text{Number of cubic centimeters of filtrate X assay of filtrate.}}{\text{Number of grams of solids}}$$

The gold entering the ball mill in the grinding solution equals

$$\frac{\text{Per cent of moisture in ball-mill discharge}}{\text{Per cent of solids in ball-mill discharge}}$$

multiplied by the assay of the storage solution. The gold value of the plant heads is the undissolved gold plus the dissolved gold in the ball-mill discharge minus the gold entering the ball mill in grinding solution per ton of ore. The gold value multiplied by the number of tons gives the amount of gold entering the plant for the day.

This rather involved procedure has been found necessary for two reasons:- First, it is impossible to get a consistent, reliable sample of the ore entering the mill without expensive sampling equipment to reduce a very large sample. Second, evaporating a pulp containing gold in solution causes segregation of the gold in the pulp and on the dish, making the sample entirely unreliable.

Tailings Sample.- The tailings sample is taken by hand, quartered down wet to about 600 grams of solids, and filtered and washed with water to make about 600 c.c. of filtrate. Then the

$$\frac{\text{Number of cubic centimeters of filtrate x assay of filtrate}}{\text{grams of solids}}$$

equals dissolved gold loss per ton, which added to the assay of the solids, gives the total loss of gold per ton. It is assumed that during each day the quantity of solids discharged equals that entering the mill as ore. This is a false assumption, but over a month's time it approximates the truth; and the number of tons milled multiplied by the grade of the tailings approximates the loss of gold in the tailings. The difference between the gold entering the plant and the gold discharged is estimated as the recovery. Over a period of years this estimate agrees almost exactly with bullion recovery. On a month's run it may vary as much as 15 per cent, partly due to varying amounts of gold trapped in classifiers, launders, and in the tube-mill lining.

Press Recovery Estimate.- A drip sample is taken from the pipe entering the gold solution tank and is called "press heads." A drip sample is taken from the discharge solution from the press and is called "press tails." Press heads minus press tails multiplied by number of tons precipitated is the estimated press recovery. Once during each shift the flow of clarified solution to the gold tank is stopped for 30 minutes. The number of inches of solution precipitated out of the tank is measured, and it is assumed that the precipitation takes place at the same rate throughout the shift. From the diameter of the tank and the number of inches pumped per hour, the solution tonnage precipitated is calculated. The press recovery is calculated from number of tons precipitated multiplied by the difference between press heads and press tails. Over a period of years this checks very closely with bullion recovery. For a year it checks within 1 per cent. For one month there may be a variation of 10 per cent, partly due to gold absorption in the furnace lining.

Other Samples.- Besides the above samples a sample is regularly taken to show the dissolved gold going to the filter and a sample of the underflow and overflow of the bowl. The bowl samples are quartered down, filtered, and washed to determine the undissolved gold.

WATER SUPPLY

Mine water is used for water wash on the filter. It is first used as cooling water in the mine compressors and thus materially reduces the cost of heating the mill. One-fifth of a ton of water is used on the filter per ton of ore and just replaces the solution carried out by the discharged cake. The mine water, while rather hard, carries no harmful impurities. Water for tailings disposal amounting to about 1,000 gallons per ton of ore, is partly mine water and partly purchased from the municipal plant of Kirkland Lake.

LABOR

Seven men operate the plant, working 8-hour shifts, as follows:

	Cost per shift
Three solution men, at..	$5.00
Three ball mill men, at	4.50
One crusher man, at........	4.00
Average daily wage....	4.64

Table 1.- Screen sizes

Kirkland Lake Gold Mining Co., Ltd.

Screen	Ball-mill feed	Ball-mill discharge	Tube mill discharge	Classifier rake return	Classifier over-flow	Bowl under-flow	Bowl over-flow
Minus 2 inch ring plus 1 inch square..............	31.6	-	-	-	-	-	-
Minus 1 inch square plus ½ inch square..............	30.7	-	-	-	-	-	-
Minus ½ inch square plus 3 mesh..............	13.8	-	-	-	-	-	-
Minus 3 mesh plus 10 mesh......	13.7	2.0	-	2.5	-	-	-
Minus 10 mesh plus 20 mesh....	4.1	7.3	-	10.0	-	-	-
Minus 20 mesh plus 40 mesh....	2.5	17.5	0.5	12.5	-	-	-
Minus 40 mesh plus 65 mesh....	1.4	20.4	1.5	13.0	-	-	-
Minus 65 mesh plus 80 mesh....	.3	8.1	2.0	7.9	-	-	-
Minus 80 mesh plus 100 mesh..	.3	7.5	7.2	10.7	-	-	-
Minus 100 mesh plus 150 mesh	1.6[1]	5.1	9.7	14.1	-	-	-
Minus 150 mesh plus 200 mesh	-	4.0	17.0	10.7	32.0	47.5	1.0
Minus 200 mesh plus 300 mesh	-	28.3[1]	62.1[1]	18.6[1]	8.2	22.8	3.0
Minus 300 mesh..............	-	-	-	-	59.8	29.7	96.0

01 - Smallest screen used.

Note:- The analyses in this table are not averages but are typical of operations in the last six months of 1930.

Table 2.- Average mill assays

Kirkland Lake Gold Mining Co., Ltd. Period covered: Year of 1930

Product	Gold, dollars	Gold, ounces
Average heads	11.38	0.553
Average tailings	1.22	.0593
Pregnant solution	3.86	.1877
Barren solution	.044	.0022
Dissolved loss	.0661	.00322
Bowl overflow undissolved	2.19	.1065
Bowl underflow undissolved	4.54	.222

Percentage recovered, 89.27

Table 3.- Metallurgical data

Kirkland Lake Gold Mining Co., Ltd. Period covered: Year of 1930

Head assay		$11.38
Total ore treated	tons	52,768
Days operated		365
Hours operated per day		21.4
Operating time	per cent	89.3
Average tonnage per 24 hours		142.6
Recovery of gold per ton		$10.16
Consumption of cyanide[1]	pounds	73,000
Consumption of lime	do.	315,956
Consumption of zinc	do.	3,718
Consumption of lead nitrate	do.	520
Soluble gold in tailings per ton of ore		$0.0661
Net water consumption, gallons per ton		1,000
Ball consumption	pounds	234,377
Liner consumption	do.	26,200

1 - Aero brand 50 per cent NaCN

Table 4.- Analysis of pregnant solution

Cyanide, as KCN	0.038 per cent
Lime, CaO	.0972 do.
Thiocyanate, as KCNS	.2323 gram per liter
Ferrocyanide, as $K_4Fe(CN)_6$	Nil
Ferricyanide, as $K_6Fe(CN)_6$	Trace
Soluble sulphates, sulphur	.02025 gram per liter
Fe	.0015 do.
Ca	.0130 do.
Pb	.0043 do.
Cu	.0115 do.
Te	Nil
Zn	.032 gram per liter
Au	5.5 mgm per liter
Ag	1.06 do.

1 - Cyanide compounds are reported as potassium salts although not known to be present in that form.

Note:- Reducing power of solution: 1 c.c. requires 0.18 c.c. $\frac{N}{10}$ $KMnO_4$ under standard conditions.

Table 5.- Analysis of tailing

	Per cent
Silica	49.6
Alumina	11.85
Ferric oxide	1.52
Ferrous oxide	6.18
Manganese oxide	.13
Lime	6.8
Soda	6.0
Potash	4.44
Carbon dioxide	9.01
Titanium dioxide	.87
Phosphorus pentoxide	.13
Pyrite	1.84
Molybdenite	None
Lead	Trace
Tellurium	.003
Copper	None
Gold	.0001, or $0.80 per ton
Silver	Trace

Table 6.- Summary of costs

Kirkland Lake Gold Mining Co., Ltd. Period: 1930. Milling method: All sliming.

Ton of ore treated: 52768

Production: Gold, 25774 ounces
Silver, 3593 ounces

	Operating labor	Supervision	Power	Supplies	Reagents	Repair labor	Miscellaneous	Total
Crushing........	$0.0504	$0.01	$0.0263	$0.0401		$0.0139	$0.0043	$0.1450
Grinding........	.0866	.02	.2555	.2342		.0394	.0060	.6417
Cyanidation..	.1093	.03	.0847	.1031	$0.153	.0458	.0063	.5322
Refining........	.004	.01	.0012	.0347		.0004	.0033	.0536
Sampling and assaying....	.0091			.0064			.001	.0165
Total............	.2594	.07	.3677	.4185	.153	.0995	.0209	1.389

Note:- Water is charged as supplies.

No interest, head-office charges, taxes of amortization are included.

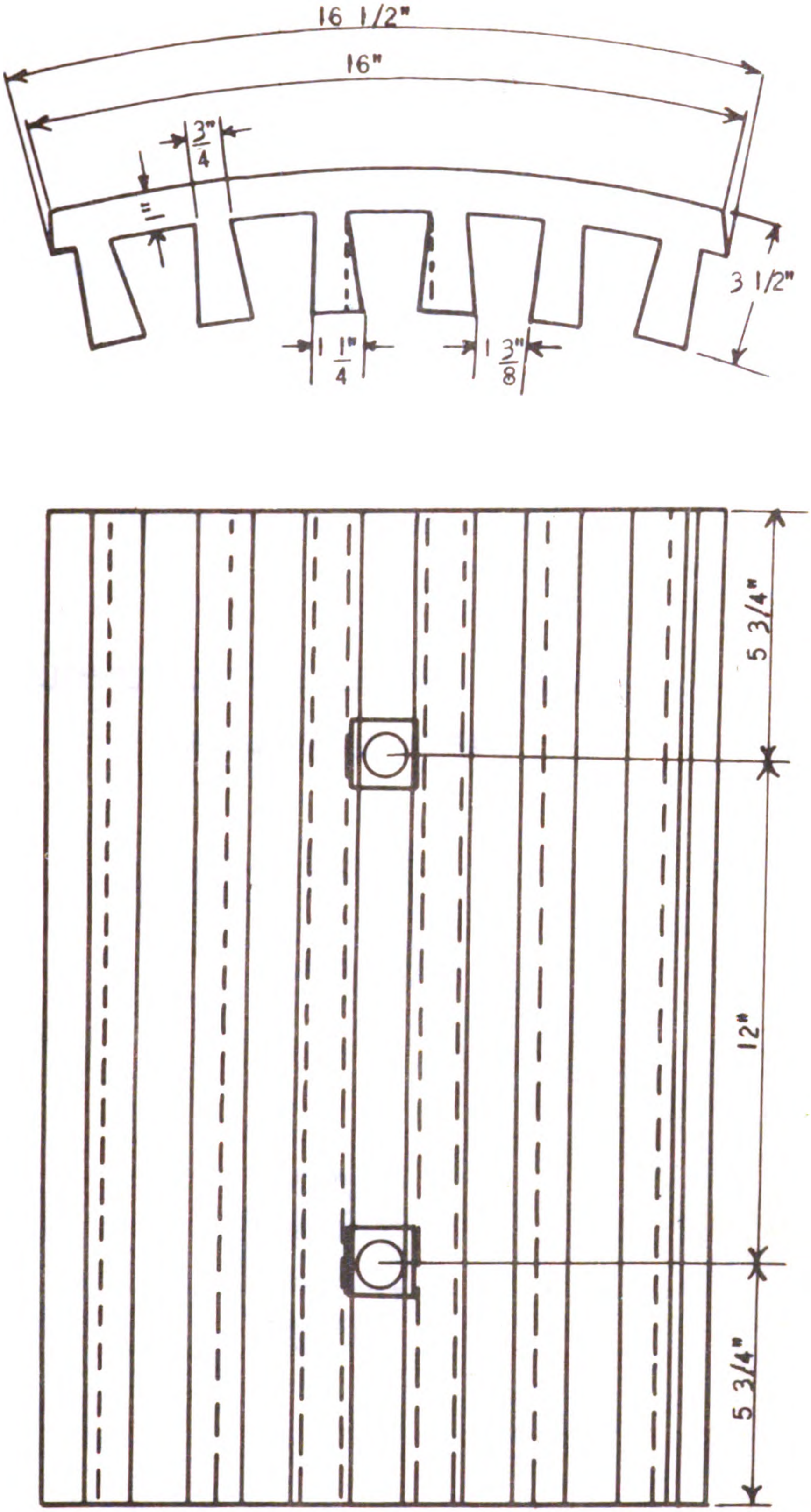

Figure 3.- Shell liner for 5 1/2 by 16 foot tube mill